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FLOTATION OF CHROMITE FROM CHROMITE ORE OBTAINED FROM BIRD RIVER, MANITOBA

P.R.A. Andrews
Mineral Processing Laboratory

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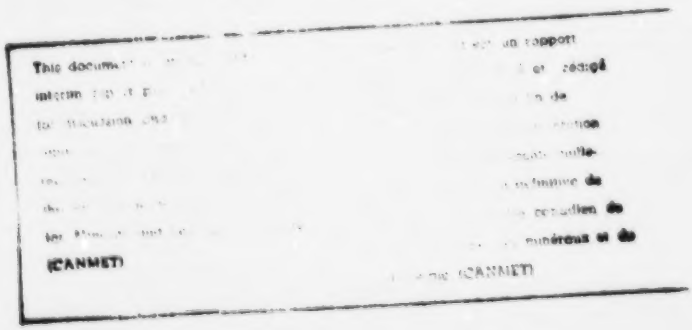

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by

P.R.A. Andrews*

ABSTRACT

A programme of batch flotation testwork has demonstrated that a low grade chromite ore from Bird River, Manitoba is best floated at pH 2.0 using Armac C (coco amine acetate) collector. A grind of approximately 100% -150 µm is necessary for liberation, and flotation selectivity is greatly enhanced by prior desliming. Rougher flotation was best conducted by stage adding collector for a total of four rougher floats. After cleaning, a concentrate analysing 21.1% Cr_2O_3 for a recovery of 93.7% was obtained from ore grading approximately 6.5% Cr_2O_3. Similar grade and recoveries were obtained during the pilot-plant evaluation.

Such selective flotation in a region of positive surface charge would imply cationic activation. It is postulated that activation could be by sulphate ion, hydrolized $FeOH^{2+}$ or CrO_4^{2-} species.

Keywords: chromite flotation, pilot-plant, activation

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* Minerals Engineer, Industrial Minerals Section, Mineral Processing Laboratory, Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources Canada, Ottawa, K1A 0G1

FLOTTATION DE LA CHROMITE CONTENUE DANS LE MINERAI DE CHROME PROVENANT DE BIRD RIVER (MANITOBA)

par

P.R.A. Andrews*

RÉSUMÉ

Un programme expérimental portant sur la flottation discontinue d'un minerai de chrome à faible teneur provenant de Bird River (Manitoba) a démontré que la flottation du minerai est plus facile à un pH de 2,0 en utilisant le collecteur Armac C (acétalamine de noix de coco). La pulvérisation de tout le minerai à une granulométrie de -150 um est nécessaire pour la libération. Un deschlammage antérieur améliore considérablement la flottation sélective. Une flottation plus rigoureuse a été réalisée par étapes en ajoutant le collecteur lors de quatre essais de dégrossissage. Après le nettoyage, on a obtenu un concentré dont la teneur en Cr_2O_3 était de 21,1 % et le taux de récupération de 93,7 %, à partir du minerai qui contenait environ 6,5 % de Cr_2O_3. Un concentré et un taux de récupération semblables ont été obtenus lors de l'évaluation à l'installation expérimentale.

Une flottation sélective de ce genre dans une région de charge positive de surface devrait comprendre l'activation cationique. On a émis l'hypothèse que la flottation pourrait être activée par un ion de sulfate, du $FeOH^{2+}$ hydrolysé ou les espèces CrO_4^{2-}.

Mots-clé : Flottation de la chromite, installation expérimentale; activation.

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*Ingénieur (Traitement des minéraux) Section des minéraux industriels, Laboratoire de traitement des minéraux, Centre canadien de la technologie des minéraux et de l'énergie, Énergie, Mines et Ressources Canada, Ottawa, K1A 0G1

INTRODUCTION

In an earlier study (Andrews and Jackman, 1988) of the beneficiation of chromite ore from Bird River, Manitoba, several methods of beneficiation were explored. These included ore sorting, heavy-media separation, gravity concentration and flotation. Flotation was used to recover chromite from the gravity concentration tailings, but due to the high efficiency of gravity concentration, very little chromite reported to the gravity concentration tailings. Since flotation was not fully evaluated, a laboratory study was undertaken to determine the optimum parameters of chromite flotation.

SCOPE OF STUDY

In the previous study, preliminary testwork conditions for chromite flotation were pH 2.3 using Armac C (coco amine acetate) collector. Selectivity also was improved by desliming prior to flotation. Whether these parameters are the optimum conditions for chromite flotation was the subject of this investigation. Optimisation of parameters was established first on a laboratory scale followed by confirmation at the pilot-plant level.

LABORATORY TESTWORK

Head sample

One hundred and fifty kg of pre-crushed -6 mm ore was received in the Industrial Minerals Section in July 1986. The ore was further size reduced to 630 µm using a combination of jaw and roll crushing; the crushed product was riffled into 1 kg bags.

A 1 kg head sample was ground to -150 µm, previously determined as the liberation size, and the ground product wet screened at various sizes down to 45 µm. A portion of the -45 µm fraction was further sized down to 9 µm using the laboratory cyclosizer. Each sized product was analysed for Cr_2O_3 and a product balance is reported in Table 1.

Grinding

The grinding procedure used was to grind 1 kg of flotation feed material for 20 min at 50% solids pulp density using 13 mm and 20 mm diameter burundum ceramic grinding media. The ground product was wet screened at 150 µm and the oversize, approximately 50% by weight, ground for a further 20 min at 50% solids. This procedure was considered to grind best to liberation size without producing an excessive quantity of chromite slimes.

Flotation

Since flotation selectivity is influenced by desliming, two preliminary timed rougher float tests were conducted to verify this fact; results are reported in Table 2. A subsequent series of tests were conducted to ascertain the optimum desliming size and time. After grinding, each ground product was classified for different periods of time from 2 min to 12 min using a 25 mm Mozley hydrocyclone with a 3.2 mm vortex finder and 4.8 mm spigot discharge. The slime fraction (overflow) from each test was collected and analysed; the underflow was used as flotation feed material. Rougher flotation was carried out at pH 2.3 using stage addition of 100 to 125 g/t of Armac C collector at 30% solids, followed by two stages of cleaning. Each flotation product was analysed for Cr_2O_3 by X-ray flourescence and the product balances of each test are reported in Table 3.

The influence of pH was investigated, after desliming at optimum desliming test conditions, by varying the pH from pH 2.0 to 9.6. The same stage collector addition flotation procedure was used and product analyses and balances for each test are reported in Table 4. Reagent concentration data for all tests are reported in Table 5.

PILOT - PLANT TESTWORK

Sample preparation

Ten t of -6 mm chromite ore was crushed to pass 2 mm by a combination of crushing and screening at 2 mm, with re-crushing of the +2 mm oversize. The crushed product was stored in drums as feed material.

Pilot-plant layout

The -2 mm feed material was fed via a laboratory size feeder into a 0.6 m by 0.8 m rod mill. The rod mill discharge was pumped to a 150 µm, 0.75 m diameter Sweco screen, the screen oversize feeding a 0.3 m by 0.9 m regrind rod mill; the regrind mill discharge being returned to the Sweco screen. The Sweco screen undersize was pumped to a laboratory 25 mm diameter desliming Mozley hydrocyclone, where approximately 5% weight of the feed was removed as slime and pumped to final tailings. The cyclone underflow was further deslimed in a 1 m diameter dewatering thickener, where an additional 5% weight of feed was removed as slime in the thickener overflow; the thickener underflow being feed to flotation.

The flotation feed was conditioned in a 0.36 m^3 laboratory conditioner with H_2SO_4 to pH 2.3 and pumped to two Denver #7 flotation cells, where Armac C collector was added and a rougher concentrate obtained. The rougher tails were fed to a bank of six Denver #5 flotation cells (8.5 L each), where further addition of collector recovered a scavenger concentrate from the first three cells; the concentrate from the last three cells was returned to the roughers. The rougher and scavenger concentrates were combined and fed to a bank of three Denver #5 flotation cells, Armac C added, and a cleaner concentrate obtained; the cleaner tails being returned to the scavenger circuit. The cleaner concentrate was recleaned three times with recirculation of recleaner tailings to the previous stage; each recleaner stage consisted of three Denver #5 flotation cells.

Four test runs were conducted and summaries of grades and recoveries for each run with flotation parameter and reagent concentration data are presented in Table 6. The pilot-plant flow diagram is represented in Fig. 1.

RESULTS

Laboratory testwork

The size and chromite distribution values in Table 1 indicate that 19.6% weight of the feed can be rejected at 9 µm, which represents only 6.2% distribution loss of chromite. The -9 µm fraction is mainly serpentinized gangue low in chromite, which inhibits the selective flotation of chromite.

Evidence of this supposition is indicated in Table 2 by comparing Test 1 results, without prior desliming, to Test 2 results, deslimed at approximately 10 μm before flotation. In Test 1 chromite was observed to float only in the third timed rougher float, whereas in Test 2 chromite flotation occurred in the second timed rougher float.

The results in Table 3, where the influence of desliming time on flotation performance was assessed, indicated that flotation performance appeared to be independent of desliming time. A summary of these results is presented as follows:

		Concentrate	
Desliming time (min)	Slime % Wt	Analysis % Cr_2O_3	Distribution %
2	4.4	21.1	93.6
4	7.8	14.1	92.3
6	10.7	20.1	91.0
8	17.4	19.3	90.9
10	15.7	21.1	82.6
12	18.2	19.0	88.5

The results indicate that a short desliming time of 2 min is sufficient for selective chromite flotation, although only approximately 25% of the -10 μm slime fraction was removed. The size of the individual cyclone overflow products was not measured but previous experience has demonstrated that for ground material of the same size distribution, the desliming size is approximately 10 μm.

Employing Test 3 desliming -10 μm slime conditions, the influence of increasing pH from 2.0 to 9.6 on flotation recovery was investigated. It is clear from the results in Table 4, and the pH/flotation recovery curve in Fig. 2 that pH 2.0 is the optimum pH.

Pilot-plant testwork

The first test run was insufficiently deslimed which resulted in an excessive collector consumption of approximately 2 kg/t, although good grade and recovery of concentrate were obtained. In Run #2 the feed rate was

increased to 46.3 kg/h, and as a result of improved desliming, total collector consumption was reduced to approximately 1.5 kg/t. The grade and recovery were lower than in Run #1 because of re-circulation build up of cleaner tailings, which resulted in reducing the cleaner collector concentration and consequently grade and recovery in order to maintain a balanced circuit. The same problem was encountered in Run #3 so that in Run #4 the circuit was run for at least six hours before sampling so that re-circulating cleaner tailings would be in equilibrium. The best results were obtained in Run #4 in reduced overall collector concentration at approximately 0.6 kg/t, and improved grade and recovery at 22.3% Cr_2O_3 and 93.0% recovery.

Sampling was conducted over a one hour period at 20 min intervals, sampling time being 15 sec. Recovery was calculated using the two-product formula based on feed to flotation rather than the dry feed. Thus losses of chromite in the slimes were not accounted for, but since the estimated loss of chromite from slimes was approximately 10% it is considered that the overall metallurgy would not be adversely affected.

PHYSICO - CHEMICAL CONSIDERATIONS

Optimum flotation was obtained at pH 2 and recovery decreased steadily as the pH was increased, as demonstrated by the results in Fig. 2. Since the pzc of chromite is in the range 4.4 to 9.6 (Palmer, 1972), depending on the presence of substituted elements in the crystal lattice, flotation by cationic collectors would not normally be expected. As such, activation of chromite could explain flotation which was markedly selective.

Anionic adsorption

Divalent anions, such as sulphate, have been shown to adsorb specifically on positively charged surfaces. Shergold et al. (1968) observed that between pH 0.8 to 2.0 sulphuric acid produced an excellent flotation response for hematite with dodecyl amine collector. Similar oxide activation by sulphate has been demonstrated by Nakatsuka et al. (1970) for ilmenite, and by Andrews (1971) for cassiterite using dodecyl ammonium acetate collector. It is possible that chromite is similarly activated by sulphate when using coco amine acetate collector since sulphuric acid was

used as the pH modifier. Furthermore, chromium is known to form very stable insoluble complexes with sulphate ion (Sillén and Martell, 1964); the adsorption of a $CrSO_4^-$ species would reverse the surface charge. A similar mechanism of activation by fluoride ion was postulated by Abido, 1971, since chromium forms the very stable CrF_2^- complex. Significantly no flotation was observed when HCl was used as pH modifier which would indicate that chloride iron does not activate chromite. The stability constant for $CrCl_2^-$ reported by Sillén and Martell, 1964 indicates that $CrCl_2^-$ is much less stable than $CrSO_4^-$.

Cationic adsorption

Polyvalent metal ions also adsorb specifically on oxide surfaces by hydrolysis of a particular cation to its first hydroxy complex. Hydroxy complexes are such surface active species that they will adsorb on a positively charged surface. Fuerstenau et al. (1970) demonstrated that the hydrated ferric ion $FeOH^{2+}$ at pH 2.5 was adsorbed on positively charged alumina.

The composition of Bird River chromite head material at 6.90% Cr_2O_3, 9.44% Fe, 22.9% MgO and 8.3% Al_2O_3 (Andrews and Jackman, 1988) would imply the impure spinel variety, where Mg^{2+} is substituted for Fe^{2+}, while Fe^{3+} and Al^{3+} are frequently substituted for Cr^{3+}. Other minerals present in significant quantities are magnetite and serpentine; the former providing a source of Fe^{3+} while the latter Mg^{2+} and Al^{3+}. Equilibrium data for logarithmic concentration of hydrolized species in solution as a function of pH indicates that at pH 2.0 only $FeOH^{2+}$ would be present in significant concentration (Butler, 1964).

Another possibility is the formation and adsorption of CrO_4^{2-} species which exist at pH 2.0, so that cationic collector species could be adsorbed at CrO_4^{2-} sites. Smith and Allard, 1983 demonstrated that CrO_4^{2-} have an affinity for chromite surfaces. In the region of pH 2 the presence of CrO_4^{2-} reduced the adsorption of dodceyl sulphate ions on positively charged chromite.

CONCLUSIONS

Batch flotation testwork has demonstrated that for the system chromite - Armac C, flotation is optimum at pH 2 after grinding to approximately

100% -150 µm. The ground ore should be deslimed at approximately 5 to 10 µm, but it is not necessary to discard more than 5% by weight as slime. By stage adding 125 g/t of collector for a total of 500 g/t, a concentrate analysing 21.1% Cr_2O_3 for a recovery of 93.7% was obtained. Similar grade and recoveries were obtained during the pilot-plant evaluation.

ACKNOWLEDGMENTS

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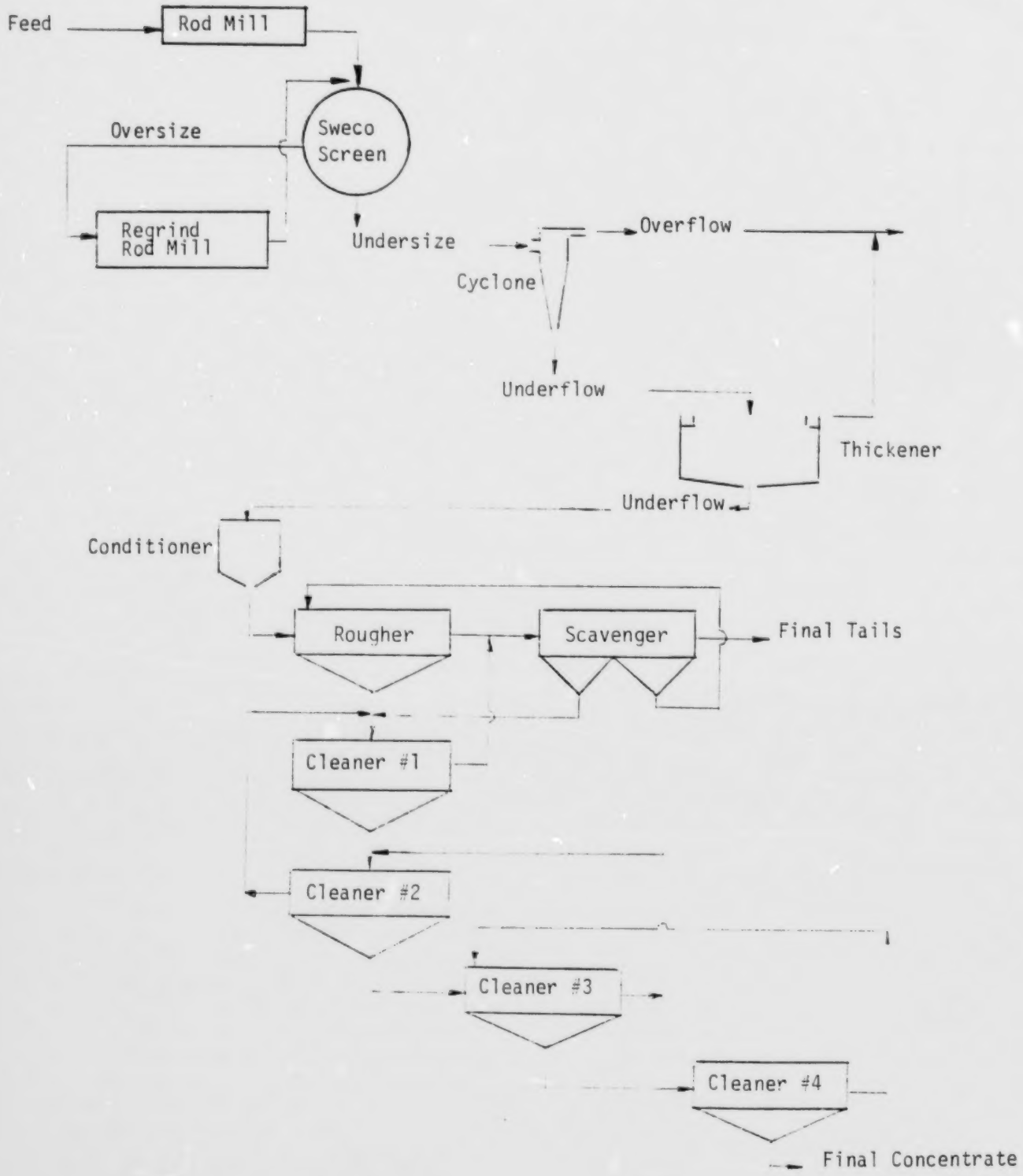

Fig. 1 - Chromite Flotation Pilot Plant Flow Diagram

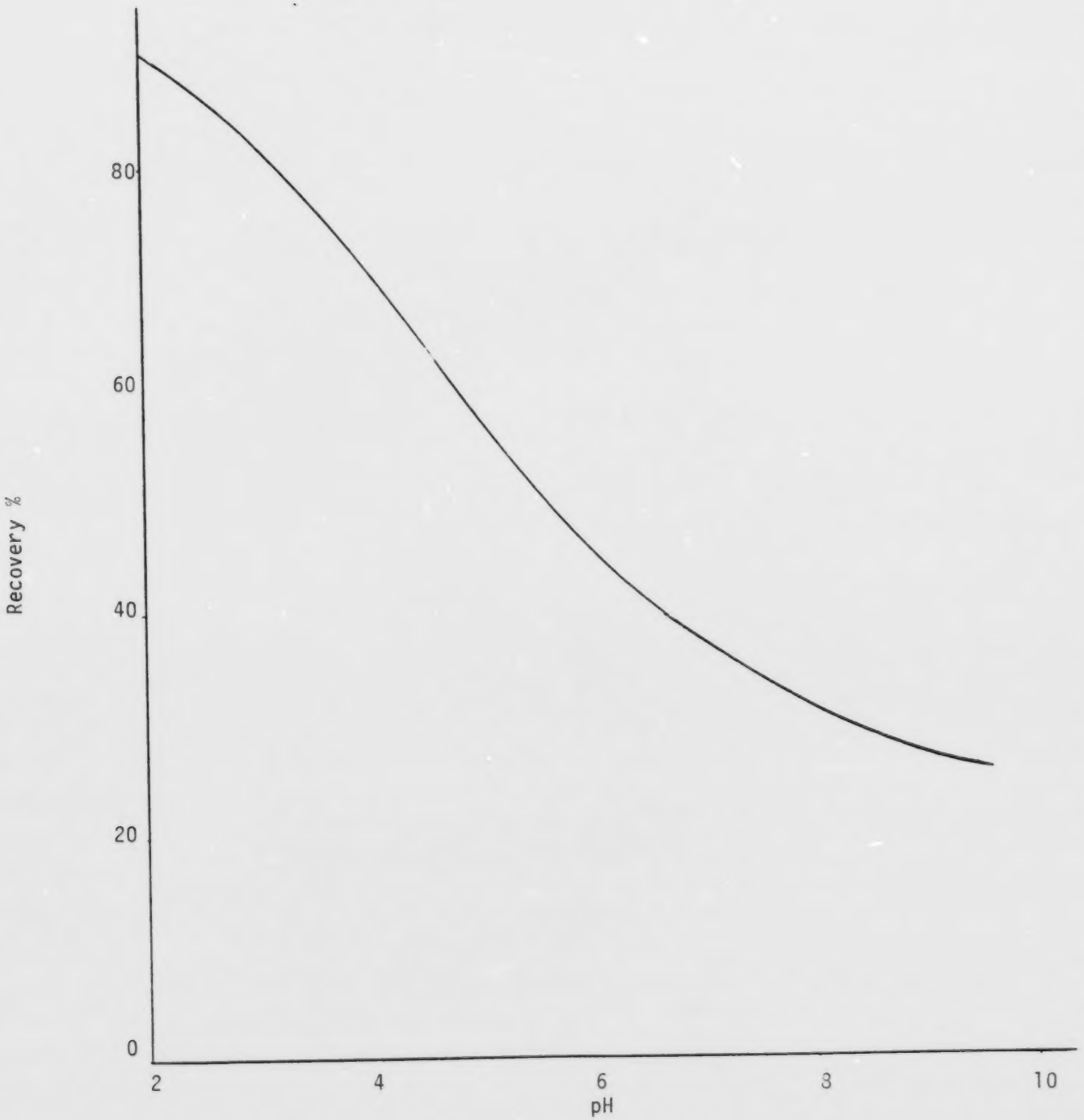

Fig. 2 - Relationship between flotation recovery and pH

Table 1. Sieve analysis and chromite distribution

Size µm	Wt %	Analysis % Cr_2O_3	Distribution % Cr_2O_3
-160 +100	7.5	6.6	8.8
-100 + 70	11.2	7.0	13.9
- 70 + 45	17.2	8.3	25.2
- 45 + 35	6.8	16.0	19.2
- 35 + 26	12.2	5.3	11.4
- 26 + 18	9.2	3.9	6.3
- 18 + 12	9.6	3.2	5.4
- 12 + 9	6.7	2.7	3.2
- 9	19.6	1.9	6.6
Head (calculated)	100.0	5.66	100.0
Head (measured)		6.05	

Table 2. Influence of desliming

Test No.	Products	Wt %	Analysis % Cr_2O_3	Distribution % Cr_2O_3
1	1st Rgh. float	3.2	2.15	1.1
	2nd Rgh. float	8.9	3.82	5.4
	3rd Rgh. float	18.6	16.5	48.8
	4th Rgh. float	12.6	15.1	30.3
	Rgh. tails	56.7	1.60	14.4
	Head (calculated)	100.0	6.29	100.0
2	Slime	18.6	1.61	4.8
	1st Rgh. float	6.4	3.16	3.3
	2nd Rgh. float	17.5	16.0	45.0
	3rd Rgh. float	14.3	17.6	41.0
	4th Rgh. float	5.6	2.92	2.6
	Rgh. tails	37.6	0.54	3.3
	Head (calculated)	100.0	6.22	100.0

Table 3. Influence of desliming time

Test No.	Products	Desliming time min	Wt %	Analysis % Cr_2O_3	Distribution % Cr_2O_3
3	Slime	2	4.4	1.68	1.1
	Final conc.		28.8	21.1	93.6
	2nd Cleaner tails		3.2	0.39	0.2
	1st Cleaner tails		17.7	0.67	1.8
	Rgh. tails		45.9	0.46	3.3
	Head (calculated)		100.0	6.49	100.0
4	Slime	4	7.8	1.78	2.3
	Final conc.		28.8	19.1	92.3
	2nd Cleaner tails		4.6	0.78	0.6
	1st Cleaner tails		12.5	0.57	1.2
	Rgh. tails		46.3	0.46	3.6
	Head (calculated)		100.0	5.96	100.0
5	Slime	6	10.7	1.80	3.1
	Final conc.		27.7	20.1	91.0
	2nd Cleaner tails		5.6	0.98	0.9
	1st Cleaner tails		11.4	0.43	0.8
	Rgh. tails		44.6	0.57	4.2
	Head (calculated)		100.0	6.12	100.0
6	Slime	8	17.4	1.95	5.5
	Final conc.		29.1	19.3	90.9
	2nd Cleaner tails		4.4	0.97	0.7
	1st Cleaner tails		10.6	0.48	0.8
	Rgh. tails		38.5	0.33	2.1
	Head (calculated)		100.0	6.18	100.0
7	Slime	10	15.7	2.07	5.4
	Final conc.		23.5	21.1	82.6
	2nd Cleaner tails		1.9	0.77	0.3
	1st Cleaner tails		11.0	0.87	1.6
	Rgh. tails		47.9	1.27	10.1
	Head (calculated)		100.0	6.00	100.0
8	Slime		18.2	1.76	5.7
	Final conc.	12	26.3	19.0	88.5
	2nd Cleaner tails		4.2	1.57	1.2
	1st Cleaner tails		9.2	0.56	0.9
	Rgh. tails		42.1	0.49	3.7
	Head (calculated)		100.0	5.64	100.1

Table 4. Influence of pH on flotation recovery

Test No.	Products	Wt %	pH	Analysis % Cr_2O_3	Distribution % Cr_2O_3
9	Slime	5.8	2.0	1.95	1.7
	Final conc.	24.2		24.15	90.2
	3rd Cleaner tails	6.4		2.95	2.9
	2nd Cleaner tails	2.6		0.98	0.4
	1st Cleaner tails	5.0		0.50	0.4
	Rgh. tails	56.0		0.50	4.4
	Head (calculated)	100.0		6.48	100.0
10	Slime	5.4	3.0	1.96	1.7
	Final conc.	23.1		22.84	82.7
	3rd Cleaner tails	3.0		1.75	0.8
	2nd Cleaner tails	7.3		1.42	1.6
	1st Cleaner tails	15.3		1.04	2.5
	Rgh. tails	45.9		1.49	10.7
	Head (calculated)	100.0		6.38	100.0
11	Slime	6.2	4.0	2.06	1.9
	Final conc.	32.1		14.55	70.9
	3rd Cleaner tails	2.7		2.53	1.0
	2nd Cleaner tails	5.1		2.45	1.9
	1st Cleaner tails	9.7		3.39	5.0
	Rgh. tails	44.2		2.88	19.3
	Head (calculated)	100.0		6.59	100.0
12	Slime	5.4	5.0	2.09	1.8
	Final conc.	29.5		11.0	51.8
	3rd Cleaner tails	3.9		3.86	2.4
	2nd Cleaner tails	6.0		3.49	3.3
	1st Cleaner tails	14.1		4.84	10.9
	Rgh. tails	41.1		4.54	29.8
	Head (calculated)	100.0		6.27	100.0
13	Slime	5.4	6.0	2.04	1.7
	Final conc.	30.3		11.2	52.0
	3rd Cleaner tails	4.4		3.28	2.2
	2nd Cleaner tails	9.7		3.46	5.1
	1st Cleaner tails	8.9		2.97	4.0
	Rgh. tails	41.3		5.53	35.0
	Head (calculated)	100.0		6.53	100.0
14	Slime	4.8	9.6	1.97	1.4
	Final conc.	16.0		10.5	25.5
	3rd Cleaner tails	5.6		4.17	3.5
	2nd Cleaner tails	16.5		6.02	15.1
	1st Cleaner tails	42.1		7.06	45.1
	Rgh. tails	15.0		4.12	9.4
	Head (calculated)	100.0		6.59	100.0

Table 5. Flotation parameter data and reagent concentrations data

Test No.	Flotation Stage	No. of Stages	Cond. time min	Float time min	pH	Armac C g/t	M.I.B.C. g/t
1	rougher	4	20	20	2.0	351	253
2	rougher	4	20	16	2.0	368	265
3	rougher	4	16	15	2.0	502	86
	cleaner	2	8	4	2.0	126	43
4	rougher	4	16	12	2.0	460	87
	cleaner	2	8	5	2.0	109	44
5	rougher	4	16	12	2.0	458	92
	cleaner	2	8	5	2.0	115	46
6	rougher	4	16	16	2.0	534	107
	cleaner	2	8	5	2.0	134	53
7	rougher	4	16	10	2.0	434	130
	cleaner	2	8	7	1.9	109	52
8	rougher	4	16	13	2.0	402	104
	cleaner	2	8	6	1.9	101	52
9	rougher	4	16	11	2.0	729	125
	cleaner	3	12	10	2.0	547	94
10	rougher	4	16	21	3.0	510	88
	cleaner	3	12	8	3.0	383	66
11	rougher	4	16	17	4.0	503	86
	cleaner	3	12	10	4.0	377	-
12	rougher	4	16	19	5.0	487	84
	cleaner	3	12	10	5.0	365	63
13	rougher	4	16	17	6.0	486	42
	cleaner	3	12	15	6.0	364	-
14	rougher	4	16	21	9.9	466	80
	cleaner	3	12	13	9.4	350	60

Table 6. Pilot-plant testwork

Run No.	Feed Rate kg/h	Pulp Density % s	pH	Coll. Conc.		Concentrate	
				Rgh g/t	Clean g/t	Grade %	Rec. %
1	38.4	19.5	1.9	1314	727	21.0	89.6
2	46.3	16.0	2.3	1141	408	19.3	83.7
3	47.4	13.5	2.0	830	454	15.2	80.6
4	47.2	21.5	1.6	195	434	22.3	93.0